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SOVIET NONFERROUS METALLURGY

NO. 10

SELECTED TRANSLATIONS

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FOREWORD

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Introduction

This is a serial publication containing selected translations on nonferrous metallurgy in the Soviet Union. This report contains translations on the subjects listed in the table of contents below.

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1. The Possibility of Heavy-Suspension Concentration of Ores from the Nerchinsk Deposits

Following is a translation of an article written by S. V. Bessonov and I. M. Kulikov in Izvestiya Vysshikh Uchebnykh Zavedeniy. Tsvetnaya Metallurgiya (News of Higher Educational Institutions. Nonferrous Metallurgy) No. 6, June 1959, pages 47-51; CSO: 3001-N/7 (10).7

Concentration in heavy suspension ["dense medium"] is being applied on an ever broader scale in the practice of the concentration of minerals.

As applied to polymetal ores, this method is utilized for primary concentration prior to flotation. In this case, concentration in heavy suspensions makes it possible to increase the productivity of concentrator plants and to reduce the expenditures on concentration caused by the isolation and dumping of a considerable amount of gangue during a comparatively coarse crushing of ore. This is attested by the experience of a number of foreign plants (Meskot, Khalkin, Ledvil, Eagle Picher, and others in the United States; Sullivan in Canada; Mechernich, Ramsbek, and others in West Germany) which employ heavy-suspension concentration for the primary concentration of lead, zinc and lead-zinc ores and old-dump tailings.

In the Soviet Union the heavy-suspension concentration of polymetal ores has been tested in a few plants. Thus, e. g., positive results were yielded by semi-industrial tests of a sulfidic polymetal ore at the Kumyshkansk Ore Mine and by laboratory tests of the polymetal ores of a number of other deposits as well.

It is of interest to investigate the possibility of applying heavy-suspension concentration to the lead and lead-zinc ores of Eastern Transbaykal', whose territory is dotted with a considerable number of small deposits. Many of these deposits, not infrequently having rich ores, are not being worked at present in view of the unprofitableness of constructing concentrator plants there and the uneconomicality of the haulage of their ore to the existing concentrator plants of the Nerchinsk Ore Administration, because of the considerable intervening distances.

Heavy-suspension concentration could, in our opinion, make it possible in a number of cases to conduct preliminary concentration on small stationary or movable installations.

servicing one or several adjacent ore mines. The ore concentrated at these installations could be delivered for final concentration to the existing concentrator plants or to central concentrator plants especially built to service groups of ore mines. If this process proves to be effective for the ores currently concentrated in the plants of the Nerchinsk Ore Administration, then its introduction will make it possible to increase the productivity of enterprises without any major capital outlays and to improve their operating indexes.

The studies of the ores of certain Nerchinsk deposits conducted in 1957-1958 at the Problems Laboratory of Mineral Concentration of the Irkutsk Mining and Metallurgical Institute, included, in addition to the other methods of concentration, an investigation of the possibility of the primary concentration of these ores in heavy suspensions. Investigated were the lead ore of the Pervo-Ignatovskiy Deposit and the lead-zinc ores of the Blagodatskiy and Kamenskiy deposits.

The present article recapitulates in brief the results of the researches in the concentration of the above-named ores in heavy suspensions.*

The bulk of the lead (81 percent) in the ore of the Pervo-Ignatovskiy Deposit is represented by galenite and boulangerite, and the oxidated lead minerals are represented by cerussite and crocoite. Sphalerite, chalcopryrite and covellite are not present in industrially exploitable amounts. The metallic minerals are represented by considerable amounts of arsenopyrite, pyrite, and certain others. The nonmetallic minerals are represented mostly by quartz (predominating amount), calcite, and feldspar, and sericite and chlorite are encountered as well. The nature of the mineralization is veined-impregnated. In addition to the fairly large segregations of galenite and boulangerite there also occurs a fine impregnation of these and other minerals and complex mutual intergrowth of the metallic minerals.

The lead-zinc ore of the Blagodatskiy Deposit is characterized by a high content of pyrite and zinc and by a more complex and closer mutual intergrowth of metallic minerals. Lead is represented chiefly by galenite and boulangerite, and zinc -- by sphalerite. This ore contains arsenopyrite and inconsiderable amounts of chalcopryrite and other metallic minerals. The bulk of the nonmetallic minerals is represented by calcite, dolomite and quartz.

The mixed sulfidic-oxidized lead-zinc ore of the Kamenskiy Deposit is characterized by a low content of lead

*

The experimental part of the study was done in collaboration with Engineer S. V. Shcherbakova

(1.25 percent) and zinc (1.02 percent). The zinc contained in the form of oxidized minerals reaches 29 percent of the total zinc content of the ore. Boulangerite and arsenopyrite are absent. The nonmetallic minerals are represented mainly by quartz, plagioclases, feldspars, biotite, chlorite, sericite, and others. This ore lacks a large segregations of metallic minerals: a major part of galenite and sphalerite occurs in the form of minute dispersed impregnation in the vein mass and in the form of complex intergrowths and intergrowths with other metallic minerals as well.

Heavy-suspension concentration was applied to an ore ground to -25, -15 and -10 mm. During every experiment the ore suspension was first washed on a screen with holes having a diameter of three mm for the purpose of eliminating fines. The ϕ 3 mm class was concentrated in heavy suspension. The suspension substance used was a finely ground and re-cleaned gravity concentrate isolated from the high-grade part of the ore of the Pervo-Ignatovskiy Deposit and having the following composition: 65.87 percent of Pb and 3.59-8.62 percent of Zn; specific gravity -- 5.60-6.37. Granulometric characteristic (in percent):

ϕ 0.074 mm	--	28-14
-0.074 ϕ 0.044 mm	--	26-10
-0.044 mm	--	46-76.

The concentration was conducted in cylindrical vessels, with manual stirring. After careful stirring and stratification of the ore (in the course of 10 seconds) the fraction that had floated up to the top was skimmed off with a mesh scoop; the settling-out heavy fraction was periodically removed, in measure with its accumulation, from the vessel. After the ending of the experiment every fraction was washed with water on a screen with 1-mm holes.

Experiments with the concentration of the ores of the Pervo-Ignatovskiy Deposit in a suspension with a specific gravity of 2.7 demonstrated the possibility of separating tailings with a lower content of lead (0.34-0.46 percent) than in experiments with the direct flotation of ore (0.74 percent). The greatest yield of dump tailings (38.2 percent) is obtained when crushing the ore to -25 mm; at a finer crushing the yield of tailings declines because of the increase in the amount of the then forming fines (class ϕ 3 mm). The losses of lead in dump tailings are virtually identical at crushing within the range of 25-10 mm.

In the suspension with specific gravity of 2.8 the greatest yield of tailings (44.3 percent) is obtained also when crushing the ore to -25 mm, and the zinc content in

Table 1

Results of the Heavy-Suspension Concentration of Ore

Density of Suspension 3 in grams/cm ³	Product of Concentration	Yield %	Content Pb, %	Recovery Pb, %
2.7	Concentrate Tailings Screenings (class -3mm)	46.7	15.53	75.8
		38.2	0.34	1.3
		15.1	14.93	22.9
	Raw Ore	100.0	9.74	100.0
2.8	Concentrate Tailings Screenings (class -3 mm)	40.5	16.37	72.9
		44.8	0.43	2.1
		15.2	14.93	25.0
	Raw Ore	100.0	9.09	100.0

the tailings is approximately the same (0.43 percent). The results of the experiments in the concentration of an ore crushed to -25 mm are cited in Table 1.

As can be seen from Table 1, the ore of the Pervo-Ignatovskiy Deposit concentrates satisfactorily in heavy suspensions.

Tests of the ore of the Blagodatskiy Deposit also had yielded positive results, although in this case the relative yield of tailings was smaller than in the case of the ore of the Pervo-Ignatovskiy Deposit. This last circumstance is partly to be explained by the higher content of metallic minerals, especially pyrite, in the Blagodatskiy ore, and by the finer impregnation of minerals as well. In a suspension with a density of 2.8 grams/cm³ the yield of tailings amounts to 19.4-22.4 percent, and these tailings contain 0.26-0.41 percent of Pb and 0.30-0.59 percent of Zn. In this connection, commensurately, the losses of metals in tailings do not exceed 2.1-2.4 and 1.0-1.8 percent, respectively. The content of lead in the flotation tailings amounts to 0.45-0.55 percent, and of zinc -- 0.70-0.80 percent. Concentration in heavy suspension proceeds most effectively when the ore is crushed to -25 mm and the suspension has a density of 2.8 grams/cm³.

During studies of the ore of the Kamenskiy Deposit it was established that it concentrates poorly in heavy suspensions. The yield of tailings ranges then from 19.2 to 44.2 percent, and their content of metals reaches 0.46-0.66 percent of Pb (at a 1.25-percent of Pb content in ore) and 0.66-0.70 percent of Zn (at a 1.02-percent content of Zn in ore), whereas the tailings yielded by direct flotation contain 0.16-0.18 percent of Pb and 0.10-0.12 percent Zn.

For a fuller assessment of the process of concentration in heavy suspensions, as applied to the afore-named ores, experiments with the flotation of the heavy fraction from the ore of the Pervo-Ignatovskiy Deposit and fines (class -3 mm) were conducted, and in this connection both products were floated both separately and jointly.

Joint flotation yielded better indexes.

The heavy fraction and the -3 mm-class in every experiment were combined and crushed in a roll crusher to -2 mm, whereupon the material was ground in a rod mill at a solid:liquid:stoke ratio of 1:0.5:6 with a 71.4-percent content of the -0.074 mm class (optimal grinding size of raw ore prior to flotation. The following flotation agents and conditions were applied: (butyl) xanthogenate, 100 grams/ton; cresol, 30 grams/ton; and time of flotation, 12 minutes; control flotation -- sodium sulfide, 300 grams/ton;

xanthogenate, 100 grams/ton; cresol, 22.5 grams/ton; and time of flotation, 18 minutes.

By way of example let us cite the indexes obtained during the flotation of the heavy fraction yielded by concentration of ore crushed to -25 mm jointly with screenings (-3 mm class) in a suspension with a specific gravity of 2.7.

Table 2

Technological Indexes of Ore Concentration

Product of Concentration	Yield in %	Content of Pb, in %	Recovery of Pb, in %
Concentrate	17.2	48.74	90.6
Intermediate Product	4.6	5.92	3.1
Aggregate Froth Product	21.8	39.54	93.7
Aggregate Tailings	78.2	0.74	6.3
Raw Ore	100.0	9.20	100.0

The results of the conducted experiments attest to the possibility of obtaining comparatively high technological indexes when concentrating ore by the combined system incorporating the process of concentration in a heavy suspension and subsequent flotation of the heavy fraction jointly with the screenings (-3 mm class). The recovery of lead and primary froth product containing 46.00-39.00 percent of metal amounts to 91.7-94.4 percent. The content of lead in the aggregate tailings of flotation and suspension-concentration does not exceed the content of that metal in the tailings of the direct flotation of raw ore (approximately 0.7 percent). The consumption of flotation agents per ton of processed ore in the above experiments proved to be 15-25 percent lower than in the experiments with direct flotation.

Conclusions

1. Laboratory investigations have established in principle the feasibility of the heavy-suspension concentration of the ores of two deposits (Pervo-Ignatovskiy and

Blagodatskiy) of the Nerchinsk Group; the third of the tested ores (Kamenskiy Deposit) is not suitable for such concentration.

2. As a result of the concentration of ores it is possible to segregate as much as 22-44 percent of wastes with a lower metal content than that of the tailings of direct flotation.

3. Experiments with the flotation of the concentrated product (of the Pervo-Ignatovskiy Deposit) jointly with screenings have yielded highly satisfactory results as to the recovery of lead, and they have involved a 15-25 percent lower consumption of flotation agents than in the case of the flotation of the raw ore.

The positive results yielded by two ores point to the necessity of investigating the applications of the given process to the ores of the other deposits in the Nerchinsk Group, as many of them have a material composition and character of mineralization that are close to those of the tested ores.

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2. Recovery of Selenium from the Slimes of Sulfuric-Acid Plants

Following is a translation of an article written by V. D. Ponomarev, Ye. A. Buketov, and G. A. Kononenko in Izvestiya Vysshikh Uchebnykh Zavedeniy. Tsvetnaya Metallurgiya (News of Higher Educational Institutions. Nonferrous Metallurgy), No. 6, June 1959, pages 47-51; CSO: 3001-N/7 (11).7

Selenium, which at present is of major importance in view of its applications in the newest fields of engineering, is obtained mainly from the anode slimes yielded by the electrolytic refining of copper. Another source of selenium is the slimes of the sulfuric-acid plants which utilize the [gases of the] roasting of pyrite and other sulfidic concentrates. Hitherto, however, the recovery of selenium from the selenium-bearing slimes of sulfuric-acid plants has been inadequate, and moreover only the richer slimes containing dozens of percent of that metal (Bibl. 1) are thus processed; the slimes containing from one to several percent of selenium are not utilized, and thus a large part of the selenium is forfeited. At the same time, the needs of the national economy for selenium dictate the necessity of recovering selenium from the poor slimes also.

The study whose results are described below had for its purpose the outlining of a rational path of recovering selenium from the slimes of sulfuric-acid plants operating on the roasting gases of zinc production. Such slimes consist mainly of zinc sulfate and have the following composition (in percent): 1.1 Se; 52.0 Pb; 0.7 Hg; 9.15 SiO₂; 0.9 CaO; 27.2 SO₄; 8.95 miscellaneous.

The selenium in the slime is in its elementary state, in the red variety, and at 100°C it changes over to the gray variety. When viewed through a monocular magnifying glass, the particles of the selenium are clearly discernible; they are only slightly bound to the other particles of the slime. The granulometric composition of the slime was as follows:

φ0.35 mm	--	0.6 percent	
φ0.30 mm	--	7.65	"
φ0.21 mm	--	4.6	"
φ0.147 "	--	6.7	"
φ0.1 "	--	4.55	"
-0.1 "	--	75.9	"

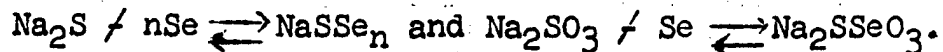
According to published data (Bibl. 2, 3), selenium can in principle be recovered from the slimes by the following methods:

1. Calcining the slime in an air current and driving off the selenium in the form of selenious anhydride, or calcining the slime without access of air, and driving off metallic selenium.
2. Treatment of the acid slime pulp with a powerful oxidizing agent so as to pass over the selenium into the solution in the form of H_2SeO_3 .
3. Leaching of slime by Na_2S or Na_2SO_3 solutions.
4. Leaching of slime by NaCl solutions; in this connection, PbSO_4 passes over into the solution while the residue is enriched with selenium.
5. Flotation of selenium from slime.

A study of the conditions of the treatment of the acid slime pulp with a powerful oxidizing agent (e. g., permanganate) demonstrated the inexpediency of employing this method in view of the poor filtrability of the pulp, considerable consumption of oxidizing agent, and lengthy duration of the process of the reduction of selenium by sulfur dioxide from the solution containing the oxidizing agent.

We have investigated in sufficient detail three methods of concentrating selenium: passing of selenium into sulfidic or sulfitic solutions, leaching of PbSO_4 from slime by NaCl solutions, and obtainment of slime concentrate by flotation.

The leaching of Se from selenium-containing slimes is based on reversible reactions:



The selection of these solvents is to be explained by the fact that they act selectively, although it is possible that a part of SiO_2 from the slimes will pass over into the Na_2S solution.

Investigations* showed that the extraction of selenium into the solution when leaching the slimes with Na_2S solutions does not exceed 27.0 percent, whereas if the leaching is done with Na_2SO_3 solutions then that extraction rises and ranges from 40 to 72 percent. It was impossible to establish the precise parameters of the optimal regime, be-

*Experiments conducted by S. L. Kabul'nikova

cause the solutions were unstable and the results, unreproducible.

Lead sulfate is the principal component of the investigated slime. When PbSO_4 is leached with NaCl solutions, lead chloride passes into the solution. The solubility of lead chloride hinges on the concentration of NaCl, and on its temperature (Table 1).

It seemed expedient to test the method of leaching slimes by NaCl solutions, so as to concentrate selenium in the leaching residues. In addition, mercury concentrated in these residues, and this made it possible to obtain a mercury-selenium concentrate.

The methods of investigation consisted in that 10 to 25 grams of washed and dried slime were leached in a flask with a stirrer and a reflux condenser at a temperature close to boiling point. A NaCl solution (290 grams/liter) close to saturation point was used in every case.

The leaching experiments were intended to determine the optimal duration of leaching and the optimal liquid/solid ratio for obtaining the maximally selenium- and mercury-rich concentrate, and to establish, as well, the possibility of counter-current leaching (for a more complete utilization of the lixiviating ability of solutions) and regeneration of NaCl.

To determine the optimal duration of leaching, 25 grams of slime were treated with 150 milliliters of solution. After leaching the pulp was subjected to filtration in a Buechner funnel. The residue was washed with 25 milliliters of water. PbCl_2 started to segregate from the filtered solution in measure with its cooling. The leaching residue was dried at 90-100°C and weighed. Analyses showed absence of Se and Hg in the solutions.

The results of the experiments are summarized in Table 2, from which it can be seen that 40 minutes is to be regarded as the optimal duration of leaching, because any further prolongation of leaching does not result in an appreciable increase in the extraction of PbSO_4 into the solution. Subsequently, all experiments were conducted at a 40-minute duration of leaching.

To establish the optimal liquid/solid ratio, a series of experiments using a NaCl solution (220 milliliters) was conducted for various initial liquid/solid ratios ("liquid" is constructed as volume of solution in milliliters).

As can be seen from the cited data (Table 3), the greatest reduction in the weight of slime is achieved at a liquid/solid ratio higher than 20:1. When $L/S = 22.1$ L/S -- liquid/solid ratio, 10 grams of slime was reduced

Table 1

Solubility of $PbCl_2$ (grams/liter) in NaCl Solutions

NaCl Concentration in grams/liter	Temperature in °C									
	0	20	40	60	80	100	140	180	220	300
13	7	3	1	0	0	0	1	3	5	13
50	11	3	1	3	4	3	7	10	12	35
100	21	1	1	13	12	15	31	53	47	55

Table 2

The Degree of Reduction of Weight of Slime during Prolonged Leaching

Leaching Time, in minutes	20	30	40	50	60
Weight of Leaching Residue, in grams	173	161	163	152	152
Decrease in Weight of Slime, in percent	30.8	35.2	33.3	39.0	39.1

Table 3
The Degree of Reduction of Weight of Slime as a
Function of the L/S Ratio

L/S	9:1	12:1	14:1	16:1	18:1	20:1	22:1	25:1
Original Weight of Slime (grams)	12.0	18.32	15.7	13.75	12.22	11.0	10.0	8.8
Weight of Residue (grams)	10.28	7.33	5.19	4.25	3.22	2.4	1.60	1.56
Degree of Reduction (times)	2.1	2.5	2.8	3.2	3.8	4.58	6.25	6.70

Table 4

	Primary Leaching	Secondary Leaching	Tertiary Leaching
Weight of Leaching Residue, in grams	38.0	32.4	19.1
Weight of Leached Sulfate, in grams	1.0	2.6	14.3

6.25 times in weight, and the residue contained 7.1 percent Se and 4.3 percent Hg, i. e., a sufficiently selenium- and mercury-rich concentrate was obtained. A reduction in the L/S ratio results in an increase in the amount of the sulfate passing into the same volume of solution. For instance, when $L/S = 10$, the amount of sulfate passing into the solution is 38-39 percent greater than when $L/S = 20-22:1$. Consequently, it is more expedient to employ counter-current leaching in order to utilize more fully the dissolving ability of NaCl. On the other hand, during the dissolution of $PbSO_4$, SO_4^{2-} and Pb^{2+} ions pass into the solution. Obviously, if we utilize the considerable decrease in the solubility of $PbCl_2$ at a drop in temperature and precipitate it after every leaching operation, it is possible to use the mother liquor for secondary leaching.

To clarify the pertinency of these postulates, a series of experiments in counter-current leaching was conducted. After preliminary experiments, the triple counter-current mode was adopted. The materials used in the primary leaching were fresh slime and the liquor of secondary leaching; the materials used in secondary leaching were the cake of primary leaching and the liquor of tertiary leaching; the materials used in tertiary leaching were fresh NaCl solution and the cake of secondary leaching. After each leaching the liquor was filtered and the cake was washed with 25 milliliters of water. The volume of the leaching liquor amounted to 250 milliliters, and the initial weight of the slime was 40 grams. The filtered solution was cooled and left standing for 24 hours. After the separation of the crystallizing $PbCl_2$ by filtration, the solution was evaporated to the volume of 250 milliliters and re-used for the next stage of leaching.

As a result of such a triple counter-current mode, and because of the crystallization of $PbCl_2$, it is possible to use one and the same solution to leach additionally a definite amount of sulfate from the slime. As can be seen from Table 4, when triple leaching based on the counter-current principle was applied with 250 milliliters of solution to 40 grams of suspension, 20.9 grams of $PbSO_4$ were passed into the solution, whereas during single leaching only 14.8 grams of $PbSO_4$ are passed from the same amount of suspension to the same volume of solution. Thus, during triple leaching the degree of extraction increases by 41.2 percent. Here also, however, the consumption of NaCl remains rather high. Unless the NaCl is regenerated, this process lacks the promise of economicality.

The dissolving ability of the NaCl solution decreases mainly because of the sulfate ion, and therefore it is ex-

expedient to precipitate that ion, e. g., by calcium chloride.

The experiments were conducted in the following manner. A 10-gram suspension of slime was treated with 220 milliliters of NaCl solution. The leaching residue was filtered, washed with 20 milliliters of water, and the solution was stored for 24 hours to crystallize $PbCl_2$; after the separation of $PbCl_2$ the solution was evaporated until its volume shrank to 220 milliliters, and $CaSO_4$ was precipitated from it, during boiling, with a $CaCl_2$ solution concentration: 209 grams/liter. After the filtration of $CaSO_4$ the liquor was re-used for the next stage of leaching. A fresh portion of 10 grams of slime was used in every leaching stage or operation.

The first series of experiments in this cycle showed that the amount of the precipitant should not be greater than required by stoichiometry, because otherwise the excess calcium ions remaining in the solution precipitate during the next leaching stage or operation and downgrade the leaching cake. This circumstance was taken into account when conducting the subsequent experiments, whose results, (Table 5) indicate that the content of lead remains within the limits established during the first series of experiments (except for experiment No. 2), and that the cake is downgraded mainly by the $CaSO_4$ precipitating during the leaching.

The solubility of $CaSO_4$ in hot water is 0.5 gram/liter, and consequently it might be assumed that it should precipitate nearly completely from the solution. However, in our experiments, less than half of the calcium sulfate precipitated a major part of that sulfate had remained in the solution and, upon the next leaching, at an increase in sulfate-ion concentration, it precipitated and thus downgraded the cake.

It might be assumed that an equilibrium was not reached (the precipitation was conducted for 20 minutes), and thus $CaSO_4$ did not precipitate completely. This assumption was tested during the next series of experiments, which was conducted by somewhat different methods: to wit, the sulfate was precipitated not separately but jointly with $PbCl_2$. For this purpose, $CaCl_2$ was introduced before the crystallization of lead, and the pulp was boiled for 30 minutes and left standing for 24 hours. In this connection it was assumed that the formation of $PbCl_2$ crystals will be conducive to a more complete precipitation of $CaSO_4$. The results of this series of experiments did not corroborate the above-mentioned assumption.

It follows from the foregoing that the regeneration of NaCl by means of $CaCl_2$ is in principle feasible but the use of such regenerated solutions for leaching yields a concentrate that is downgraded (diluted) by calcium sulfate, be-

Table 5
Data on the Leaching of Slimes by Regenerated NaCl Solutions

Number of Experiment	Weight of Leaching Residue in grams	Content of CaO in the Residue, percent	Content of Pb in the Residue, percent	Amount of CaCl ₂ Solution, in milliliters
1	1.6	4.48	3.04	12
2	3.9	17.16	8.48	12
3	3.6	—	6.35	13
4	3.7	21.5	3.86	13
5	3.7	20.4	3.92	11
6	3.9	20.35	5.74	13
7	3.9	19.35	5.94	12
8	3.9	21.3	5.5	12
9	4.0	19.83	6.35	14
10	3.9	24.9	3.08	14
11	4.7	53.5	2.63	14

cause of the comparatively high solubility of CaSO_4 in chloridic solutions.

Investigations of the flotation of selenium from sulfuric-acid slimes were conducted by Professors S. P. Aleksandrov and F. N. Belash (Bibl. 4). They established that the use of kerosene and aldol as reagents yields a 35-37 percent Se slime concentrate from comparatively selenium-poor (three or four percent Se) slimes. Moreover, there exists a Japanese patent (Bibl. 5) recommending the use of pine oil (Dyupon B) as frothing agent and sodium dithiophosphate as promoter.

Our experiments in flotation were conducted both with and without reagents, inasmuch as organic compounds were found to be present in the slime. The experiments were conducted in a flotation machine with a capacity of 200 milliliters, at a L/S ratio of 4.5:1 and at a stirring time of 10 minutes. During the drying of the washed slime, as noted before, the red selenium changes over to the gray modification. The results of the experiments with the flotation of gray selenium from dried slime proved to be unsatisfactory.

In the course of studies of the raw slime (in which selenium is present in the form of red modification), the investigations were centered on the duration of flotation at a pulp pH of about 2.6 and at 18°C , without the addition of flotation reagents. In all experiments the weight of the slime suspension amounted to 45.2 grams. The results of these experiments are presented in Table 6.

Table 6

Results of the Flotation of Selenium Without Reagents

No. of Experiment	Duration of Flotation in minutes	Weight of Concentrate in grams	Weight of Tailings in grams	Se Content in Concentrate, in percent	Se Content in Tailings in percent	Recovery of Se in percent
1	30	1.2	36.1	16.8	0.49	37.0
2	40	2.5	35.1	16.9	0.16	77.0
3	50	2.8	35.5	17.0	0.14	87.0

This was followed by a series of experiments in which sodium dithiophosphate (aerofloat) was added to the pulp.

The optimal consumption of that reagent varies from 220 to 250 grams/ton; an increase in its consumption to 350 grams/ton reduces the yield of the concentrate and reduces its content of selenium.

The experiments with the flotation of selenium by dithiophosphate (240 grams/ton) were conducted at pH=2.4. Their results (Table 7) indicate that the prolongation of flotation to more than 40 minutes increases the yield of the concentrate but downgrades it, because this does not increase the recovery of selenium into the concentrate.

Table 7

Results of the Flotation of Selenium with an Aerofloat

Number of Experiment	Duration of Flotation in minutes	Content of Se in Concentrate, in %	Recovery of Se in %
1	20	21.22	66.0
2	30	21.21	77.0
3	40	16.33	87.0
4	50	16.13	87.0
5	60	9.7	37.0

As for the experiments to determine the effect of acidity of the medium, these were conducted at a flotation time of 30 minutes, and they showed that when the pH value exceeds 3 then the recovery of selenium into the concentrate drops by one-half.

It was established that mercury also passes into concentrate, together with selenium. At a Se content of 16-21 percent in the concentrates, the Hg content varied correspondingly within the limits of 10-13.5 percent.

The slimes in which selenium was present in the form of the gray modification were subjected to experiments. These experiments dealt with a 50-gram slime suspension at a time, at L/S = 4:1 and at room temperature (Table 8).

Table 9 cites the results of the effect of an aerofloat at a leaching time of 30 minutes.

As can be seen from the data cited, the gray selenium from these slimes is poorly floated, and its

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Table 8

Results of the Flotation of Gray Selenium Without
Flotation Reagent

Duration of Flotation in minutes	Weight of Concentrate in grams	Se Content in Concentrate, in percent	Recovery of Se, in percent
10	0.40	11.78	9.4
20	0.55	12.05	13.26
30	0.68	11.78	16.0
40	1.00	12.3	24.6
50	1.70	10.5	35.7
60	1.05	10.0	21.0

Table 9

Relationship Between Recovery of Gray Selenium into
Concentrate and the Consumption of Aerofloat

Consumption of Reagent per ton of Slime, in grams	Weight of Concentrate in grams	Se Content in Concentrate in percent	Recovery of Se, in percent
100	2.55	6.9	35.20
150	2.15	9.46	36.36
200	3.5	5.3	37.1
250	2.05	—	—
300	2.5	6.8	34.0
350	3.8	3.0	22.8

content in the concentrate varies from five to 12 percent, and its recovery into the concentrate does not exceed 35-37 percent. The poor floatability of gray selenium, which we had noted before, may possibly be attributed to the circumstance that amorphous red selenium has a larger specific surface than gray selenium.

Conclusions

1. Studies of the leaching of selenium from low-grade sulfuric-acid slimes containing 1.10 percent Se and 0.7 percent Hg by solutions of sodium sulfide and sulfite have shown that this method cannot be recommended, in view of the low degree of its recovery of selenium into the solution and its high consumption of dissolving reagents.

2. Studies of the obtainment of mercury-selenium concentrate by leaching $PbSO_4$ from slimes by sodium chloride solutions have shown that this method can be used to obtain concentrates containing seven percent Se and four percent Hg, and it ensures a complete recovery of these metals into the concentrate; however, the practical application of this method would not be easy in view of its high consumption of sodium chloride and the difficulties in regenerating that reagent.

3. Experiments with the flotation of selenium from slimes showed that selenium, probably because of the presence of organic compounds in the slimes, can be floated without using flotation agents; however, the addition of an aerofloat improves the flotation conditions somewhat. In the concentrates, the Se content varies from 16 to 21 percent, and the Hg content, from 10 to 13.5 percent.

4. The optimal conditions for flotation are: pH = 2 (not over 3), 40 to 50 minutes (30 to 40 minutes on introducing, into the pulp, an aerofloat in the amount of 220-250 grams/ton of slime), and L/S = 3-5.

5. Red selenium is more easily floated than gray, which may possibly be explained by the greater specific surface of the amorphous red selenium; therefore, in sulfuric-acid plants it is necessary to avoid conditions under which red selenium changes over to gray selenium.

6. The retreatment of the obtained slime concentrates of flotation so as to extract from them selenium and mercury presents no special difficulties: such concentrates are blended with soda ash and calcined for several hours at a temperature of about $700^{\circ}C$; in this connection, the mercury is driven off and trapped in the condensers, the calcining residue is leached with water, and the selenium passes into soda solutions from which it is precipitated by the conventional method of reduction with sulfur dioxide after acidification of the solution.

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3. Obtaining Tellurous Concentrate from the Melts of Alkali Lead-Refining

Following is a translation of an article written by V. S. Lovchikov and B. M. Lipshitz in Izvestiya Vysshikh Uchebnykh Zavedeniy. Tsvetnaya Metallurgiya (News of Higher Educational Institutions. Nonferrous Metallurgy), No. 6, June 1959, pages 93-98; CSO: 3001-N/7 (12).⁷

During the alkali refining of lead so as to eliminate arsenic, tin and antimony, a part of the tellurium passes into the melts. At the Chimkent Lead Plant the content of tellurium in melts reaches 0.34 percent (Bibl. 1). The hydro-metallurgical retreatment of alkali melts is accompanied by the distribution of tellurium among all products, as attested by the data that we obtained (Table 1).

Table 1

Content of Tellurium in Products of the Retreatment of the Melts of the Alkali Refining of Lead

Product	Content of Tellurium	Recovery of Tellurium Into Products of the Retreatment of 100 tons of Melts, in kg
Alkali Solution of Pulp Obtained During Granulation of Melts	0.11-0.75 grams/liter	18.6-135
Solution After Repulping of Bulk Cake	0.07-0.05 grams/liter	12.8-91.5
Condensate from "Rapid" Apparatuses	none	--
Condensate from Evaporating Vats	none	--
Sodium Antimonate	0.26-0.42%	92.6-149.5
Calcium Stannate	0.11%	0.3
Calcium Arsenate*	0.22%	--
Metallic Antimony	0.08-0.3	11.6-43.3
Slag of the Reduction Smelting of Sodium Antimonate	0.39-1.06%	54.6-142.2

*At another domestic plant the content of tellurium in calcium arsenate ranges from 0.2 to 0.27 percent, which in terms of

100 tons of retreated melts amounts to 28.2 to 38.1 kg of Te.

The presence of tellurium in alkali solutions stems from the solubility of telluric and tellurous acids in sodium salts (Bibl. 2, 3). The presence of tellurium in sodium antimonate and calcium stannate is attributable to the poor solubility of the tellurites of magnesium, barium and lead in alkali solutions (Bibl. 4).

The data in Table 1 shows that tellurium is contained in all the products of the retreatment of melts except for the condensates obtained upon evaporating strong alkali solutions.

The alkali solution of the pulp obtained during the granulation of melts and sodium antimonate are the two substances of the greatest interest to the recovery of tellurium from melts. The tellurium isolated from that alkali solution will pass into bulk cake and subsequently, when certain conditions are observed, it will remain in sodium antimonate. As a result, the other products of the retreatment of melts will not contain tellurium.*

Isolation of Tellurium from the Alkali Solution of the Pulp Obtained During the Granulation of Melts

To remove tellurium from the alkali solution, it is possible to use the antimony obtained from the reduction smelting of sodium antimonate. Compared with tellurium, antimony has an electronegative potential (Bibl. 5, 6) and it cements tellurium from the solution. In this connection, arsenic is not displaced by antimony, because it forms a stable complex ion AsO_4^{3-} , and the solution lacks any other ions that might also be cemented by antimony.

The granulation of melts is accompanied by the formation of strong alkali solutions containing about 26 percent NaOH. The isolation of tellurium from these solutions was conducted by us with metallic antimony in the form of grains with sizes of -1.2/0.5 mm. The use of finely ground antimony powder does not yield a positive effect because in this case the oxidation of the antimony and the formation of soluble sodium antimonites are faster than the process of the cementation of tellurium.

Experiments with the isolation of tellurium from alkali solutions were conducted in a glass cylinder with a

*This study was done with the collaboration of M. N. Chepik, an engineer at the Ust'-Kamenogorsk Lead-Zinc combine.

mechanical stirrer. The cylinder was filled with one liter of the pulp obtained at the Chimkent Lead Plant during the granulation of an alkali melt containing 0.3 percent Te. The liquid phase of the pulp contained 1.1 grams of Te per liter. In addition to tellurium, selenium (0.4 gram/liter) was detected in the liquid phase of the pulp.

The pulp was heated to 95°C, the stirrer was switched on, and then five grams of metallic antimony were added. After definite time intervals, samples of the solution were collected for chemical analysis. In the course of the experiments the volume and temperature of the pulp were maintained constant.

The averaged data on the experiments are presented in the graph below. After an eight-hour cementation the content of tellurium in the solution declined from 1.1 to 0.06 gram/liter, i. e., 94 percent of the tellurium had passed over to the solid phase.

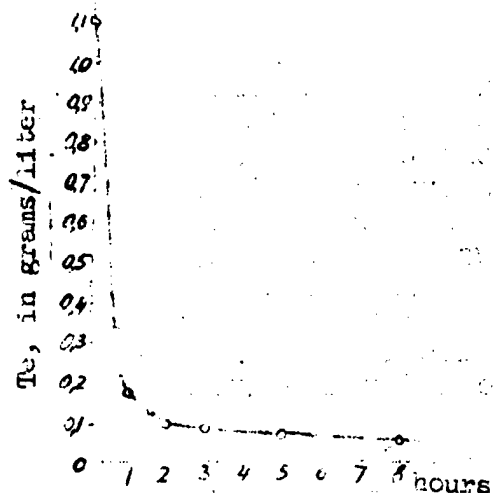
In contrast with tellurium, under the above conditions, selenium is virtually not cemented at all by metallic antimony.

From solutions containing 10-12 percent NaOH, tellurium is satisfactorily cemented by an aggressive antimony powder. When the amount of antimony used is of a weight quadruple to the weight of the tellurium in the solution, the cementation ends within 20 minutes and nearly all of the tellurium is eliminated from the solution. A double amount of antimony by weight does not eliminate all tellurium from the solution, and instead it yields a sponge containing up to 60 percent Te.

According to technical requirements, the cake obtained during the filtration of a pulp containing a strong alkali solution is to be repulped with water. In the process of repulping of the cake, the cemented tellurium again changes over to the liquid phase despite the presence of coarse-grained metallic antimony, which accounts for its dissolution in alkali solutions.

In order to obviate the dissolution of cemented tellurium, the pulp obtained from the repulping of bulk cake was enriched with an aggressive metallic-antimony powder obtained by the following method.

Sodium and antimonate in a mutual ratio of 4:1 were charged into an aluminum crucible, stirred until the formation of a homogeneous mass, and then inserted, together with the crucible, into an electric furnace and heated there at 300-400°C until brown vapors ceased to be emitted. The heated mass was cooled, crushed in a mortar, and calcined at 550-600°C for 20-30 minutes. The 100 grams of sodium antimonate thus treated yield 75 grams of an antimony powder



Rate of the Cementation of
Tellurium From a Strong Al-
kali Solution by Metallic
Antimony

containing 62 percent Sb.

In the experiments with the repulping of bulk cake the consumption of the antimony powder amounted to five grams per liter of pulp, at L/S = 5:1. The pulp was blended with the antimony powder by a mechanical stirrer at 30°C for six hours, whereupon a sample of the solution was collected and analyzed for tellurium.

The results of the experiments showed that in the presence of the antimony powder no dissolution of the cemented tellurium is observed. All tellurium concentrates in sodium antimonate. The cementation of tellurium by zinc powder occurs more quickly, but it is not expedient to use zinc powder for this purpose, because this results in increasing the zinc content in the antimonate and polluting with zinc the alkali solution.

Isolation of Tellurium from Sodium Antimonate

Sodium antimonate contains 45 percent Sb and serves as a raw material for obtaining metallic antimony and for the side recovery of tellurium.

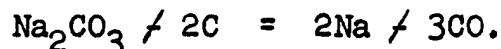
The best results in obtaining the maximum amount of metallic antimony (approximately 90 percent) are provided by smelting a charge containing 80 percent sodium antimonate, 10 percent charcoal and 10 percent sodium chloride.

To clarify the behavior of tellurium during the reduction smelting of antimonate, a series of experiments was carried out by the following method. The charge containing sodium antimonate (with 0.38 percent Te), charcoal, and 10-percent sodium chloride was moisturized to eight percent H_2O , stirred, placed in an alundum crucible and melted in a gas hearth at $900^{\circ}C$.

The investigations were centered on the effect of charcoal and temperature on the distribution of tellurium between metallic antimony and slag.

The results of the experiments (Table 2) showed that at a 15-percent and higher content of charcoal in the charge, the amount of tellurium in the metallic antimony decreases drastically and the concentration of metallic sodium increases. An analogous picture is observed when the degree of grinding of the charcoal is increased.

The presence of sodium in the antimony is to be explained (Bibl. 7) by the interaction between the soda forming during the melting of sodium antimonate and the carbon, according to the following reaction:



Sodium forms with tellurium intermetallic compounds Na_3Te , Na_2Te and $NaTe$, the last-named of which remains stable until $953^{\circ}C$ (Bibl. 8) and passes into slag during the melting of the charge. It is this property of tellurium that explains also the decrease in its concentration in antimony with increasing saturation of the latter by metallic sodium.

The effect of temperature on the Te content in the antimony obtained from the melting of sodium antimonate with 10 percent of charcoal in the -0.2 mm mesh and with 10 percent NaCl is illustrated in Table 3, from which it follows that with rising temperature the content of tellurium in metallic antimony drops until only traces are left at $950^{\circ}C$. This attests to a virtually total passage of the tellurium into slag, dust and gases. Observations of the process of melting of the charge showed a very feeble formation of vapors of metals and dust.

When such a process is industrially introduced, it is recommended that the tellurium in the form of trapped dust should be returned to the melted charge of sodium antimonate so as to ensure a more complete recovery of tellurium

Table 2

Effect of Charcoal on the Content of Tellurium in the Metallic Antimony
Obtained from the Smelting of Sodium Antimonate

Content of Charcoal in Charge, percent	Grain Size of Charcoal in mm	Recovery of Metallic Antimony in percent	Content, in Metallic Antimony, in %			
			Te	Na	Pb	As
5	-0.2	46.0	0.53	none	1.7	-
7.5	-0.2	52.0	-	traces	1.32	-
15	-0.2	95.4	0.03	0.13	2.11	0.8
25	-0.2	96.0	Traces	0.27	-	0.7
10	-0.2	94.0	0.48	traces	2.11	0.8
10	-1.2 + 0.2	94.3	0.42	-	2.11	-
10	-0.2 + 0.1	95.0	0.20	0.05	2.11	-
10	-0.1	95.0	0.15	0.07	2.11	-

Table 3

Effect of Melting Temperature on Te Content of Antimony

Temperature in °C	Recovery of Metallic Antimony, in %	Content, within Antimony, in %			
		Te	Na	Pb	As
750	90	0.53	None	1.9	0.8
850	94	0.48	Traces	2.11	0.8
950	92	Traces	0.25	2.11	0.7

in the slag.

Recovery of Tellurium from Slags

Experiments with recovering tellurium from slags were conducted in the following manner. A glass cylinder was partly filled with water and then charged with 250 grams of slag containing 0.75 to 1.2 percent Te in grains of -3 mm size. The resulting pulp was stirred for two hours at 90°C and thereupon filtered. The filtrate was analyzed for the content of Te, Na₂CO₃, NaOH, and NaCl. The solid residue was dried, weighed and assayed for Te.

The obtained results of experiments are summarized in Table 4, from which it is seen that within the investigated range of L/S ratios the leaching of slag with hot water is accompanied by a virtually total passage of tellurium into solid residue in which its amount exceed three percent. Such a product can serve as raw material for obtaining tellurium. Upon the cementation of the latter from the liquid phase of the pulp obtained from the granulation of alkali melts, the content of tellurium in the solid residue of leaching will be higher.

The solution obtained from the leaching of slags contains Na₂CO₃, NaOH and NaCl. The re-treatment of the solid residues washed on the filter with hot water does not yield solutions containing the above-mentioned compounds, which attests to the complete recovery of these compounds from slags during their primary leaching.

To obtain more concentrated solutions, it is expedient to conduct the leaching of slag at a L/S ratio of 1:3. In this case, the liquid phase will contain 44.7 percent Na₂CO₃.

Table 4

Results of Experiments in Leaching Slags with Hot Water

L/S Ratio	Amount of Solid Residue, in percent by weight of Slag	Content of Te	
		In Solid Residue in percent	In Solution
1:3	25	4.2	Traces
1:4	24	3.1	None
1:6	25	3.4	"

5.25 percent NaOH, 8.5 percent NaCl, and 1.2 percent Na₂S. Such a solution, after it is cleaned from sulfur by a lead-containing alkali reagent, can be included into the scheme of the retreatment of the melts of the alkali refining of lead.

Conclusions

1. In the course of the currently practiced hydro-metallurgical retreatment of the melts of the alkali refining of lead, the tellurium is distributed among all the resulting products except for the condensates of evaporators. The principal amount of the tellurium passes into the liquid phase of the pulp from the granulation of alkali melts, into the solution from the repulping of bulk cake, and into sodium antimonate.

2. From the liquid phase of the pulp obtained from the granulation of the melts of the alkali refining of lead, it is expedient to cement tellurium by metallic antimony, because this simplifies the technology of its extraction from the melts in view of its virtually total passage into sodium antimonate.

3. The distribution of tellurium among the products of the reduction smelting of sodium antimonate hinges on the consumption of charcoal, degree of crushing of charcoal, and temperature.

The optimal condition for passing tellurium into slag is the smelting of sodium antimonate in combination with 10-15 percent charcoal in the -0.2 mm size and with 10 percent NaCl, at 950°C, when tellurium completely concentrates in the slag and is no longer contained in metallic antimony.

4. The leaching of the resulting slags with water is accompanied by the formation of a solid residue into which passes all the tellurium. The solid residue contains

over three percent Te and can be utilized as the raw material for obtaining tellurium. The cementation of tellurium from the liquid phase of the pulp obtained from the granulation of alkali melts will make it possible to increase its content in the solid residue of the leaching of slags. At present, studies of the recovery of tellurium from that product are in progress, and their results will be published later.

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4. Recovery of Precious Metals by Anion-Exchange Resins from the Waste and Processing Solutions of Copper Electrolytic Enterprises

Following is a translation of an article written by A. B. Davankov, V. M. Laufer, S. V. Pakitin, L. G. Levian, and A. I. Chernobay in *Izvestiya Vysshikh Uchebnykh Zavedeniy. Tsvetnaya Metallurgiya* (News of Higher Educational Institutions. Nonferrous Metallurgy), No. 6, June 59, pages 134-141; CSO:3001-N/7 (13).⁷

One of comparatively new ranges of applications of ionites is the hydrometallurgy of nonferrous and, especially, precious metals, in which they have acquired of late a vital importance as an effective means of recovering gold, silver, platinum, and other metals, from industrial wastes (Bibl. 1). The continued and broader use of ionites will make it possible to reduce substantially the losses of precious metals in the liquid wastes and discards of the galvanic shops of jewelry enterprises (Bibl. 2), refining plants, mints, mirror and photographic film plants, powder-metallurgy enterprises, etc. In the immediate future ionites should find application in the recovering of gold and other precious metals from cyanous (Bibl. 5), hydrochloric-acid and iodinic solutions (Bibl. 6), and, in addition, from the liquid wastes of gold-recovering enterprises, ore mines, concentrator plants, and hydrometallurgical shops. An attempt has been made to apply ionites in recovering gold from sea water (Bibl. 7).

Ionites may be of special importance to the copper-electrolysis and lead-zinc plants, where they can be used not only for the total recovery of metals from waste liquors but also, and primarily, for removing slimes and other impurities from the electrolytes, which ensures a normal and stoppageless performance of electrolytic baths and a qualitatively good deposition of metals on cathodes.

The coagulation, absorption and segregation of the suspended particles of gold, silver, platinum, and other precious metals, from aqueous dispersions by means of ion-exchange resins has for a number of years been successfully employed under industrial conditions in the jewelry enterprises of the USSR (Bibl. 8).

The principle of the coagulation of negatively charged metal particles from aqueous dispersions by means of anion-exchange resins is based on the neutralization of these

charges by positive charges carried by nearly stationary polyvalent resin cations.

It is this principle that was utilized in the present study which was intended to isolate precious metals from the waste liquors and processing solutions of an Ural copper electrolysis plant. At that plant, precious metals are encountered in processing solutions basically in the form of suspensions of slime whose fine particles are capable of passing through very dense filters.

The elimination of slimes from the acid solutions of copper sulfate by means of ionites is of special importance in itself, because this also results in a high degree of purification of the sulfuric-acid copper, which ensures a longer performance of electrolytic baths and the yield of high-grade copper.

The technical characteristics of the waste liquors of cuproelectrolytic solution are cited in Table 1, from which it can be seen that the highest content of gold (47.2 milligrams/m³) and silver (510 milligrams/m³) is displayed by the waste liquors proceeding for cementation. Qualitative tests of waste liquors and processing solutions by an acetic-acid solution of benzidine displayed negative results as to the presence of ions of precious metals, which attests that in these liquors precious metals are present in the form of metallic suspensions.

Coagulation-adsorption methods under static and dynamic conditions were used to remove these suspensions.

In the first case, 500 milliliters of a solution with pH = 1.5-2.0 were treated, while intensively stirred for 15 minutes, with anion-exchange resins N-0, AN-1, AN-9, AN-2F, and AN-2FG, taken in the amount of 0.2 to 2.0 percent of the volume of the treated liquor. After it was allowed to settle and clarify, the solution was decanted and the next portion of fresh liquor was poured into the reaction vessel. This was repeated several times, in which connection one and the same amount of resin was used to treat from three to 20 liters of solution as depending on its composition.

During the first experiments the tested liquor was divided into two parts. One part was concentrated by evaporation, and the other -- treated with anion-exchange resin. An assay of the ash residue obtained from the burning of the resin and the dry residue obtained from the concentration of the control part of the liquor by evaporation established identical content of precious metals in both these residues. This had corroborated a virtually total recovery of Au and Ag from the tested processing liquors under static conditions. The concentration of Cu, H₂SO₄, and pH of the liquors remained virtually unchanged after treatment with ionites.

Table 1

Analyses of Shop Liquid Wastes

Liquid Wastes From	From 25 Jan 50 to 5 Feb. 1957				From 5 Feb to 15 Feb. 1957				From 15 Feb to 25 Feb. 1957			
	Cu, g/l	H ₂ SO ₄ , g/l	Au, mg/m ³	Ag, mg/m ³	Cu, g/l	H ₂ SO ₄ , g/l	Au, mg/m ³	Ag, mg/m ³	Cu, g/l	H ₂ SO ₄ , g/l	Au, mg/m ³	Ag, mg/m ³
Slime Shop	0.0032	none	5.0	50.0	0.017	none	108	136.9	0.045	0.31	7.2	221.8
Sulfate Shop	1.1	21.1	0.15	0.61	0.01	---	5.0	50.7	0.015	0.17	1.2	163
Electrolysis Shop	0.10	0.70	3.3	46.6	0.10	0.54	10.0	140.0	0.033	0.62	5.0	19.50
Copper Powder Shop	0.036	0.35	20.0	310.0	0.32	1.60	4.3	87.1	0.01	0.31	14.0	165.0
Laundry Shop -- Before Centrifugation	0.047	1.05	36.6	510.0	0.075	0.46	95.0	115.0	0.1	2.21	47.2	469.0
Laundry Shop -- After Centrifugation	0.010	0.58	3.3	163.3	0.01	0.15	15.3	183.6	0.01	0.5	21.3	381.7

Analogous results were obtained when conducting experiments under dynamic conditions, i. e., during the filtration of processing solutions at the rate of 15-25 milliliters/minute through a glass tube with a 30-mm diameter which was previously loaded with 40 grams of AN-1 anion-exchange resin in carbonate form.

It follows from the results, cited in Table 2, that metallic particles of gold and silver are completely recovered by ionites from diluted waste liquors proceeding for cementation. In this connection, the consumption of ion-exchange resins under static conditions did not exceed 0.3-0.5 percent of the volume of the treated liquor. Similar results with regard to the recovery of precious metals were obtained when treating the post-cementation waste liquors with pulveroid wastes of N-0 resin (see Table 2).

Special interest is merited by the elimination of mechanical suspensions (slimes) from the liquors of the sulfate shop. The related utilization of the coagulating and adsorbing abilities of solid polyelectrolytes insoluble in water and in organic solvents, i. e., the polyelectrolytes constituted by ion-exchange resins, is of major interest both in practice and in theory.

In accordance with the technological flow scheme adopted at the copper electrolysis plant at which the tests were conducted, the sulfate shop receives liquors from the electrolysis and slime shops. The technical characteristics of these liquors, as determined by an analysis made by central plant laboratory, are cited in Table 3, from which it follows that the liquors of the electrolysis and slime shops contain considerable amounts of precious metals which are present both in the finished products of the sulfate shop (copper sulfate and copper-nickel salts) and in the wastes sent to the copper smelting shop or to other copper smelting plants.

Commensurate experiments were accordingly conducted to test the recovery of ionites of the gold and silver from the liquors richest in these metals. Such liquors are primarily constituted by the processing solutions coming from the decopperizers of the slime shop. The experiments showed (Table 4) that the treatment of these solutions by anion-exchange results in a rapid and complete "desliming" of solutions. The elimination of slime (mechanical "impurities", including noble metals) by means of ion-exchange resins yields satisfactory results both for the cooled liquors and for the liquors heated to 50°C. It should, however, be noted that certain types of anion-exchange resins (AN-2FG and AN-9) proved to be insufficiently resistant to the action of hot liquors. When acted on by acid during heating

Table 2

Results of the Recovery of Precious Metals by Anion-Exchange Resins in Static and Dynamic Conditions From the Waste and Processing Solutions Before and After Generation

Results of the Recovery of Precious Metals by Anion-Exchange Resins in Static and Dynamic Conditions From the Waste and Processing Solutions Before and After Cementation																
Content in Initial Solution	Cu, g/l	H ₂ SO ₄ , g/l	Au, mg/m ³	Ag, mg/m ³	Volume of Resin-Treated Initial Solution in Liters	Type of Anion-Exchange Resin	Weight of Dry Residue After the Concentration-by-Evaporation of the Filtrate, Grams									
							Consumption of Anion-Exchange Resin Treatment of Solution		Recovery from Ash Residue		Residual Content in the Filtrate		Weight of Dry Residue After the Concentration-by-Evaporation of the Filtrate, Grams		Residual Content in the Filtrate	
							% of Volume of solution	Grams	% of Initial Content in Solution	milligrams	% of Initial Content in Solution	milligrams	% of Initial Content in Solution	milligrams	% of Initial Content in Solution	milligrams
	0.9	0.1	428	4438	86	N-O (Pulveroid Wastes)	40	0.46	2.0	1.88	45.6	37.90	99.9	14	232	1.0
	0.9	0.08	292	4306	89		40	0.45	2.0	2.16	100.5	39.50	103.0	57	none	none
	1.08	-	293	3900	120		50	0.43	11.0	3.52	100.0	16.97	35.3	53	none	248.3
	0.32	1.79	40	633	126	An-2F	40	0.30	1.0	0.5	99.2	7.98	100.0	11	traces	none
	0.43	0.68	40	500	90	An-2FG	50	0.55	1.5	0.35	100.0	4.50	100.0	16	none	none
	0.43	0.64	56	994	10	An-1	40	0.40	1.0	0.56	100.0	9.24	100.0	25	none	none
	0.17	0.57	80	1026	15	An-1	30	0.20	3.5	1.33	98.5	10.26	66.6	16	1.33	372.0
	0.008	1.99	28	340	20	N-O (Pulveroid Wastes)	40	0.30	0.7	0.36	100.0	0.80	100.0	86	none	none
	-	1.52	5	63	102		40	0.40	1.0	3.35	96.2	0.80	95.0	40	traces	1.6

Table 3

Results of an Assay of the Content of Gold and Silver
in Ten-Day Samples of the Liquors of the Slime and
Electrolysis Shops for the Period from 1 January to
1 May 1958

Liquor			
From Slime Shop		From Electrolysis Shop	
Au in milli- grams/ m ³	Ag in milli- grams/ m ³	Au in milli- grams/ m ³	Ag in milli- grams/ m ³
1860	15240	125	1180
898	11873	10	580
175	4953	116	1075
1046	12586	30	210
932	7404	10	115
933	11900	10.5	120
892	10869	-	-

they became subject to hydrolytic dissociation (destruction).

Attempts at purifying with ionites the strongly acid solutions sent to neutralization from the electrolysis shop proved to be unsuccessful because of the high concentration of sulfuric acid (200 grams/liter) in the solution (Table 5). For this purpose it is necessary to use ionites that are resistant to the effect of strong acids.

Major complications were also encountered when treating with anion-exchange resins the neutralized solutions yielded by oxidizers. Even a slight cooling of these solutions, with their high content of CuSO₄, led to the precipitation of crystals of copper sulfate and consequently to their losses when separating the solution from the resin residue during filtration (see Table 5).

Industrial tests of the results of the laboratory study yielded positive results. Upon the filtration of 500 liters of waste processing liquors through a vinyl plastic column filled with the N-O

Table 4

Results of the Recovery of Precious Metals by Anion-Exchange
Resins from the Processing Solutions of the Plant's
Slime and Electrolysis Shops

Content in Initial Solution				Volume of Resin- Treated Initial Solution in liters
Cu grams/liter	H ₂ SO ₄ grams/liter	Au milli- grams/m ³	Ag milli- grams/m ³	
55.4	20.27	486	9322	4.50
42.57	18.27	779	5776	4.50
26.30	25.60	561	8510	4.90
59.28	50.47	1616	17990	12.0
57.21	39.37	400	6600	4.0
32.12	21.42	408	8000	6.0
57.21	39.37	492	7436	5.0
57.21	39.37	500	8500	3.80
62.2	45.2	486	10720	5.0
62.2	45.2	486	10720	5.0
62.2	52.0	328	5364	5.0
65.0	52.0	416	6144	5.0

/Continued on Page 37/

[Continued from page 36]

Type of Anion- Exchange Resin	Consumption of Anion- Exchange Resin for Treatment of Solution		Weight of Ash Residue of the Burning of Resin, grams
	grams	% of volume of solution	
N-0 (Pulver- oid Wastes)	30	0.75	4.0
"	40	0.9	0.7
"	40	0.8	2.5
"	35	0.3	16.0
"	10	0.25	8.0
N-0 (Granular	25	0.4	2.5
AN-9	60	1.2	11.0
AN-1	40	1.0	4.5
AN-2FG	60	1.2	6.0
AN-2F	40	0.8	4.0
AN-2FG	60	1.2	19.0
AN-1	40	0.8	4.0

[Continued on Page 38]

[Continued from page 37]

Recovery from Ash Residue				Residual Content in the Filtrate		Weight of Dry Residue After the Concentration-by-Evaporation of the Filtrate, grams	
Milli-grams	Au	% of Initial Content		Au, milli-grams/m ³	Ag, milli-grams/m ³	Au, milli-grams/m ³	Ag, milli-grams/m ³
		in Solution	in Solution				
1.94	88.7	39.08	93.2	54.9	635.5		
3.43	97.8	25.0	96.1	16.8	220.4		
2.75	100.0	41.7	100.0	none	none		
19.2	99.0	214.0	99.1	16.0	156.6		
1.6	100.0	26.3	99.6	none	25		
2.45	100.0	56.0	100.0	none	none		
2.31	100.0	37.18	100.0	none	none		
2.05	107.9	32.7	101.2	none	none		
0.78	32.1	20.4	38.0	none	none		
2.4	98.8	53.6	100.0	330	6640		
1.52	92.6	24.9	92.6	6.0	none		
1.84	88.5	29.0	94.4	24.0	384		
				48.0	344		

Table 5

Results of the Recovery of Precious Metals by Anion-Exchange
Resins from the Processing Solutions of the Electrolysis
Shop Before and After Neutralization

Content in Initial Solution				Volume of Resin-Treated Initial Solution in Liters		Type of Anion-Exchange Resin	Consumption of Anion-Exchange Resin for Treatment of Solution		Weight of Ash Residue of the Burning of Resin, Grams	Recovery from Ash Residue				Weight of Dry Residue After the Concentration by Evaporation of the Filtrate, Grams	Residual Content in the Filtrate	
Cu, g/l	H ₂ SO ₄ , g/l	Au, mg/m ³	Ag, mg/m ³	Volume of Resin-Treated Initial Solution in Liters	Type of Anion-Exchange Resin	Grams	% of Volume of Solution	Weight of Ash Residue of the Burning of Resin, Grams	% of Initial Content in Solution	MLL-1, Grams	% of Initial Content in Solution	MLL-1, Grams	% of Initial Content in Solution	Weight of Dry Residue After the Concentration by Evaporation of the Filtrate, Grams	Au, mg/m ³	Ag, mg/m ³
34.0	196	300	6140	5.0	AN-2F	40	0.8	12.0	1.9	98.5	8.4	98.8	1	900	1	5012
142.1	23.0	2431	8800	3.5	AN-1	40	1.1	28.0	6.4	96.8	29.4	95.4	95.4	1100	31	400
143.0	17.0	300	3575	4.0	AN-2F	40	1.0	26.0	0.78	99.0	11.7	73.6	73.6	1000	105	1050
142.6	17.64	643	2441	3.0	AN-1	30	1.0	45.0	1.9	98.5	6.28	92.5	92.5	1300	9.6	183

granular anion-exchange resin, no gold or silver was detected in the filtrate. The liquor subjected to filtration contained, in suspended state, three milligrams of Au per m^3 and 510 milligrams of Ag per m^3 , and the solution contained 1.02 grams $CuSO_4$ per liter and 0.77 gram H_2SO_4 per liter; the rate of filtration was 30 liters per hour.

Analogous results were also obtained under static conditions during the processing of 1.8 m^3 of a copper electrolyte containing 29.84 grams Cu and 128.32 grams H_2SO_4 per liter and "polluted" with a large amount of slime. After three-hour treatment of the solution with pulveroid N-O resin taken in the amount of 0.41 percent of the volume of electrolyte (7.4 kg) and upon vigorous aero-stirring of the filtrate no gold or silver was found in the filtrate.

The filtration of the solution was conducted on a Nutsch filter at the rate of 1.5 m^3 /hour. The filtrate was completely transparent, displaying no signs of opalescence, which attested to the complete elimination of slime and a high degree of the removal of suspended gold and silver particles by ion-exchange resins from the electrolyte.

Conclusions

1. It has been established that metallic suspensions of gold and silver are completely recovered from the waste and processing solutions of copper-electrolysis plants by anion-exchange resins.

2. In view of the inconsiderable content of slime in the waste solutions it is expedient to recover precious and other metals from these solutions by the continuous procedure based on filtration through adsorption columns filled with granular anion-exchange resins.

3. Processing solutions with a high slime content which are rich in silver and gold can be treated under static conditions with pulveroid wastes of anion-exchange resins causing aggregation, coagulation and precipitation of metallic particles into the sediment. Then the electrolytic solutions prove to be very extensively purified of mechanical suspensions (slime).

4. To purify hot solutions with a high content of sulfuric acid (as much as 200 grams per liter) and copper sulfate, it is necessary to employ chemically stable ionites.

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5. The Bases for an Economic Evaluation of Mineral Deposits in a Socialist Economy

Following is a translation of an article written by S. Ya. Rachkovskiy in Izvestiya Vysshikh Uchebnykh Zavedeniy. Tsvetnaya Metallurgiya (News of Higher Educational Institutions. Non-ferrous Metallurgy), No. 6, June 1959, pages 185-194; CSO:3001-N/7 (14).7

The 21st CPSU Congress stipulated the fundamental goals confronting the country with regard to the building of the material-technical base of Communism and, in particular, providing for a huge increase in the extraction and processing of minerals. This requires a further development of the work on the prospecting for and exploration of mineral deposits, and on their industrial exploitation, both in the virgin regions of the country and in the existing enterprises.

For the purpose of assuring a steady growth of the national economy by means of raw-material mineral supplies the Seven-Year Plan of Development of the National Economy of the USSR for 1959-1965 provides for an increase of 67 percent in the over-all volume of geologic prospecting activities and for a 1.7- to two-fold expansion in the industrial capacities for the extraction of ores for ferrous and non-ferrous metallurgy (Bibl. 1).

Prospecting for new deposits and the expansion of old ore mines and construction of new ones usually require substantial funds and long periods of time. Because of the unrenewability of the mined mineral resources, a considerable part of the funds invested in mine construction cannot be utilized after the depletion of mines. Therefore, a preliminary evaluation of deposits to determine the effectiveness of their detailed exploration and mining development is of great national-economic importance.

As a result of the large scale of prospecting and exploring activities and their successful conduct, our country at present occupies the leading place in the world with regard to various types of mineral raw materials. The USSR is assured with raw materials for obtaining high-grade steel, producing lead, zinc, aluminum, etc., which are better than those of any other country (Bibl. 2).

Extensive deposits of mineral resources have been explored in all regions of the country. The large number of the discovered and generally surveyed deposits and the unfeasibility

ity and inexpediency of a concurrent detailed exploration and development of them all add particular urgency to the problem of the preliminary evaluation of deposits. Such comparative evaluations should establish which deposits, and in what order of sequence, it is most pertinent to explore first and to start construction on, in order to satisfy the needs of the country's national economy and to assure its balanced development.

Thus, the principal purpose of the evaluation of deposits in a Socialist economy is to establish the national-economic importance of deposits by determining, by the comparison method, the periods and priorities for their detailed development and industrial exploitation. In this connection it is necessary to consider the needs of the national economy and the developmental plans for both industry as a whole and individual regions of deposits, on taking into account the technical and economic indexes characterizing the effectiveness of their exploitation.

For a comparative evaluation of deposits it is not enough to use merely the geological and technical data characterizing these deposits. Geological and technical factors may affect the evaluation of a deposit in diverse and sometimes contradictory ways. Thus, e. g., a deposit may be known to contain very extensive and high-grade reserves, but these reserves occur under unfavorable conditions. As a result, its exploration and exploitation may prove to be less effective than the exploration and exploitation of other deposits whose reserves may be of lower quality but occur under conditions more favorable for their exploitation. One proof of this is the successful industrial exploitation of the low-grade porphyry ores of the Kounrad Deposit, which ores can be mined by open-strip methods; this exploitation proceeds at a more forced pace than the exploitation of certain other richer ores which occur at greater depths.

An economic evaluation of deposits makes it possible to reduce all the attendant numerous factors to more generalized synthetic indexes characterizing the national-economic effectiveness of exploitation of one or another deposit.

Criteria of Economic Effectiveness of Deposits

In a capitalist economy the economic evaluation of deposits is conducted for the purpose of determining the sales price of the ore mine or deposit, assessing the tax to be levied, uncovering the prospects for profitability when issuing stock-exchange shares, etc. For this purpose, the monetary value of a deposit is determined on the basis of the amount and rate of income (capitalist profit). In the United

States, as early as in 1887, Goskold had offered the following formula determining the monetary value of a deposit in terms of the symbols suggested by Baxter and Parks (Bibl. 3).

$$V_p = \frac{A}{r} \cdot \frac{1 - (1+r)^{-p}}{1 + r}$$

where "V_p is monetary value of the ore mine at the given moment, in dollars;
A is amount of yearly income, in dollars;
r is amortization rate of the mine, ensuring a return of the invested funds at the moment of the working of the deposit, in percent;
r' is rate of risk (rate of income), in percent;
p is number of years of exploitation of the deposit.

Analogous methods were used by K. L. Pozharitskiy (Bibl. 4) to propose an economic evaluation of deposits in a socialist economy. Although K. L. Pozharitskiy proposes the use of a single rate instead of two different rates for amortization and for risk (K. L. Pozharitskiy uses Goskold's formula as altered by Markill (Bibl. 57); fundamentally this does not change anything.

Let us assume that we have explored two deposits. The extent of the extractable reserves of the first deposit is five million tons, and that of the second, 23 million tons. The value of components recoverable from one ton of ore is identical for both deposits, but the operating expenditures at the first deposits are five rubles lower per ton, so that its exploitation will yield an income of 25 million rubles. At the other deposit the value of the recoverable components is commensurate with operating expenditures. When both deposits are evaluated on the basis of income, the second deposit is of no value. And yet, as depending on the raw materials balance and on the needs of the national economy, the larger reserves, region of site and other aspects of the second deposit may make it of greater value to the national economy than the first deposit.

The profitability of individual enterprises and branches in a Socialist economy is of great importance.

On the whole, a Socialist economy is a highly profitable one. However, it would be incorrect to substruct the economic evaluation of a deposit on the basis of its profitability alone. When a country's balance of a given metal is tight, it is quite possible (and this occurs in practice) to assign additional solid assets and manpower to obtain the metal needed by the country.

In the conditions of a Socialist economy the comparative evaluation of the economic effectiveness of a detailed exploration or exploitation of a deposit should be made on taking into account the decisions of the 15th Party Congress, which pointed to the need for basing the allocation of capital investments on the plan of the most relevant development of the Nation's economy as a whole upon the consideration of regional features (Bibl. 6), and, moreover, on the decisions of the 21st CPSU Congress, which pointed out the primary importance of the most effective orientation of capital investments, to wit, the enlargement of production capacities and increase in industrial output with the least expenditures and in the shortest possible time, coupled with a drastic increase in labor productivity and cutting of production costs (Bibl. 1).

Let us consider the significance of these directives to the economic evaluation of the effectiveness and determination of the order of sequence of exploitation of individual deposits.

Interests of the National Economy as a Whole. A Socialist national economy constitutes an economic organism evolving on the basis of expanded reproduction, for which purpose it is necessary to ensure the proper relationships and proportions in the development of individual branches of production. Proceeding from the goalanced development of the national economy, the Socialist State is interested in the production of a definite volume and variety of output. This signifies that the exploitation of deposits should be initiated according to an order of sequence which would ensure the satisfaction of the needs of the national economy within fixed deadlines.

If the national economy requires, e. g., an increase in lead output, it would be incorrect to expand only the extraction of copper and zinc, even if their production is more profitable, because in a Socialist economy the division of labor and means of production among the individual branches is regulated not by the law of value but by the needs of the national economy, by the interests of the country as a whole.

Therefore a comparative evaluation to determine the sequence of exploring activities or of exploitation of deposits may apply only to a specific metal or to metals that are

mutual substitutes.

Regional Features of the Location of Deposits. The region of location of a given deposit is of very great importance to the evaluation of the effectiveness of the exploration or exploitation of the deposit.

A deposit containing rich ores but located under unfavorable transport and economic conditions may prove to be less effective than a deposit containing poor ores but located under more favorable conditions.

Certain metallurgical plants in our country are not adequately assured with raw-material resources in their vicinity. The discovery and exploitation of deposits in the regions of these plants is of major national-economic importance and may prove to be more beneficial to the national economy -- even at relatively less favorable technical and economic indexes -- than the exploitation of richer deposits which would require the construction of special concentrator plants, new metallurgical plants, etc. Thus, e. g., the favorable evaluation of nephelines in Krasnoyarskiy Kray as a raw material for the organization of aluminum production is related to the long-range prospects for cheap electrical energy and deposits of limestones of the right quality in that region.

In a comparative evaluation of deposits it is also necessary to consider proximity to consumers as this will assure a reduction in the volume of hauls and accelerate turnover because of the shorter periods of hauls.

The goal of geographical rapprochement between industry and raw-material sources predetermines the expediency of constructing ore mines close to existing enterprises and conditions the need for prospecting for and exploring resources in the appropriate industrial regions.

When assessing a deposit it is necessary to take into account not only the effect of regional conditions on the construction and operation of enterprises but also the effect of such construction and operation on the economic development of the concerned region.

One of the fundamental principles of the geographical distribution of Socialist industry is the more rapid development of the previously backward regions of the USSR. In this sense, the development of the ore-mining industry and the exploitation of new deposits have played an exceptional role. The mastering of the coal, iron-ore, copper, polymetal, and other deposits of Kazakhstan and the construction, on their basis, of commensurate large enterprises have converted that republic into one of the most important industrial regions of the country. A similar situation has taken place

in other economic regions of the USSR also.

The development of deposits poses major requirements to the other branches of the national economy: transport, forest management, building materials industry, agriculture, etc. Therefore, the evaluation of deposits and determination of the sequence of their development should be tied to the presence of corresponding resources and to the over-all plan of economic development of a given region, and to the plans of development of other branches of the national economy as well. The strategic conditions of the region should also be considered.

Ensuring a Maximal Growth in Production Capacities and in Output

The most important task of a Socialist economy is to expand the volume of industrial output.

The effectiveness of expenditures on a given type of production is usually determined from the following formula; on the basis of volume of production per unit capital investment:

$$E = \frac{P}{K} \quad \text{or} \quad E_1 = \frac{K}{P}$$

where: P is value of yearly output in terms of wholesale prices, per ruble of capital investments

E_1 is amount of capital investments per ruble of yearly output in terms of wholesale prices

P is value of yearly output in wholesale prices, in rubles

K is scope of capital investments.

Index K should reflect the scope of expenditures on the construction of not only the ore mine but also other enterprises necessary for obtaining the finished products (concentrator plants, electric power stations, metallurgical plants, etc.). This will make it possible to determine the advantages of developing a given deposit as viewed from the standpoint of the possibility of coordinating and combining it with existing enterprises.

According to the present postulates, the expenditures on geologic-prospecting activities are

financed from the State Budget and are not included into the value of the basic funds of ore mines. We assume that when these expenditures are not regional in character but are related to the discovery of the reserves and the exploiting conditions of a specific deposit, then they should be included into the construction costs of the ore mine and taken into account when determining the effectiveness of capital investments.

In addition to the expenditures on capital construction, the index K should include also the expenses pertaining to the supplying of the future enterprise with personnel cadres, and the amount of turnover funds necessary for the operation of the enterprise.

Unit capital expenditures should be determined not only per ton of resources but also per ton of finished production, because it is important to the national economy to obtain more production per time unit.

Periods of Construction. In evaluating deposits it is greatly important to consider the duration of construction of enterprises to be built on the basis of these deposits. The more rapid the construction the more rapid will be the satisfaction of the needs of the national economy for a type of output and the more rapid the recoupment of the funds invested in the construction.

At a comparative evaluation of deposits with differing period of development it is necessary to consider the expenditures which the satisfaction of needs during the difference in development periods will incur to the national economy. In cases in which the provision of a type of output by other enterprises during the period of the difference in construction periods is in general not possible and the output in question is just then scheduled to satisfy the needs of the national economy, the time factor acquires a primary importance. At a rally of the builders of the Volga Hydroelectric Power Station imeni V. I. Lenin, while speaking of the present-day expediency of building thermal electric power stations, which can be completed within shorter periods than hydroelectric ones, even though the latter provide cheaper energy, Comrade N. S. Khrushchev emphasized that "the time factor in this business is more valuable than the direct material expenditures, because lost time cannot be compensated by any financial outlays whatsoever" (Bibl. 7).

Rise in Labor Productivity and Decline in Production Costs. The productive effect and profitability of capital expenditures are characterized by the maximal growth of output at minimal expenditures of social labor on the construction.

ion and obtainment of finished production. The principal synthetic index characterizing the effectiveness of capital expenditures on the exploration or development of a deposit is the expenditure of labor per production unit.

In the level of the socially necessary expenditures will be reflected the quality and quantity of resources, conditions of occurrence of the ore body, physicomineralogical composition of the ore, which predetermines the scheme for its dressing, economic conditions of the geographical region of the deposit, and so forth.

Labor expenditures can be most reliably considered in terms of time units. However, at the given stage, in view of the presence in the USSR of commodity production and the operation of the law of value, the consideration of the expenditures of labor on production is reflected, even though only partially, in production costs.

If the costs of the exploration or extraction of one ton of ore in deposit A are lower than the analogous costs for one ton of an ore of identical quality in deposit B, then, considering the analogousness of the structure of costs of exploration and extraction, it can be stated that the expenditures of socially necessary labor at deposit A are also lower.

For a comparative evaluation of deposits, it is impossible to restrict attention to a comparison of the costs of a ton of ore or even of a unit of the content of a metal in ore. Depending on the scheme of technological process, degree of recovery of valuable components, costs of further processing of ore, etc., the costs of the finished production may prove to be very high even when the costs of the raw material are relatively low.

Lenin's indication of the need for considering, when distributing industry geographically, "the possibility of the least expenditure of labor during the transition from the processing of raw material to all the successive stages of processing of the semifinished products until the obtainment of the finished product" (Bibl. 8) should be used unconditionally as the guidepost for evaluating deposits. Consequently it is necessary, when evaluating deposits, to determine the costs of the finished production.

Because of the conditions of occurrence of an ore body in a deposit, in the course of the working of that deposit it may become necessary to alter the mining system. The transition to deeper ore horizons may cause a change in the metal content of the ore, and in the very nature of the ore (e. g., transition from oxidized to sulfidic ores), which will affect considerably the costs of finished production. This necessitates the determination of the costs

of finished production in advance for the entire period of projected exploitation of a deposit instead of after the projected output capacity of the local ore mine has been reached.

In addition to the production costs, the profitability index can also be used as an index for the comparative evaluation of a deposit. The rate of profitability of individual deposits is determined according to the following formula:

$$H_P = \frac{C - V}{C} \cdot 100$$

where H_P is profitability rate, in percent:

- C is value of finished production recovered from one ton of ore, in rubles;
- V is amount of operating expenses per ton of raw material until the finished production is obtained, in rubles.

Also of economic interest is the profitability rate determined as the ratio of the sum of yearly income to the value of the basic funds of an enterprise. It can be expressed by the following formula:

$$\frac{(C - V) \cdot II}{K + O}$$

- where II is amount of yearly output, in tons;
- K is value of fixed assets in rubles;
- O is amount of liquid assets in rubles.

The meaning of C and V is the same as in the preceding formula.

An analysis of the prices of the production obtained from mineral resources indicates that their level deviates in a number of cases not only from value but also from production costs. Thus, the wholesale prices for the polymetal ores of the Leninogorsk and Sokol'nyy ore mines are 15 percent lower than their production costs, Mirgalisay Ore Mines -- 47 percent lower, Karabash Ore Mine -- 35 percent lower, etc. Therefore, it is more correct to determine profitability on the basis of finished production.

Correlation of Capital Investments and Production Costs

In a comparative evaluation of deposits it is possible to encounter cases in which the exploration or development of a given deposit require more capital expenditures per production unit but less operating expenditures, and vice versa. In such cases it is necessary to relate these two indexes mutually. While under capitalism preference is given to the version assuring a higher profit rate, in a Socialist society this problem should be resolved on taking into account the interests of the national economy. Thus, a vital influence on the solution of the problem may be exerted by periods of construction, region of location of deposit, etc. But in cases in which the exploitation of the deposits under evaluation ensures the interests of the national economy to an identical degree, it is necessary to correlate these indexes. In project-design practice such correlation is usually conducted by determining the periods of the recoupment of the additional capital investments through savings in operating expenditures, or by calculating the rate of return yielded by the originally invested funds.

When the latter method is used, compound or simple interest on the sum of the necessary capital investments is calculated, and the percentage of that interest is added to the operating expenditures, and this is followed by comparisons of the production costs for the individual versions.

The following formula is used to determine the periods of return of the additional capital investments:

$$t = \frac{K_2 - K_1}{C_1 - C_2}$$

where t is periods of return on additional capital investments; K_1 and K_2 are capital investments according to two versions, in rubles; and C_1 and C_2 are amounts of annual operating expenditures on the comparable production, in rubles.

In an article concerned with this problem V. V. Nozovhilov (Bibl. 9) demonstrated convincingly that the rate of the recoupment period constitutes in the quantitative sense the same thing as the calculation of the return on the originally invested funds. However, the period of the recoupment (return) of additional capital investments constitutes, in principle, something different.

Capital investments serve to conduct the mechanization of an enterprise and the perfecting of techniques and

technology, which lead to an increase in the productivity of social labor. Therefore, we are entitled to expect that of versions assuring the interests of the national economy to an identical extent, that version with the greater sum of capital investments should ensure a saving in social labor which should, within a definite period, compensate for the additional capital expenditures.

Academician S. G. Strumilin suggested that the time factor be considered in capital-investment projects by taking into account the losses ensuing from the continual depreciation of fixed assets as a result of the continuous rise in labor productivity (Bibl. 10).

The greater the investments in a given object the greater will be the detriment to the national economy as a result of their devaluation depreciation. Therefore, on comparing the versions requiring differing capital expenditures -- all other conditions remaining equal -- we are justified in assuming that the detriment stemming from the expending of additional capital investments should be compensated by reduction in operating expenditures.

The sum of the detriment (P) can be calculated from the following formula:

$$P = K \cdot O - \frac{K \cdot O_0}{(1 + r)^t}$$

where K is sum of original capital investments into fixed assets, in rubles;

O is sum of liquid assets, in rubles;

r is mean annual rise in labor productivity, in percent;

t is period of exploitation of deposit, in years.

The production costs of an output unit, on taking into account the detriment incurred by the devaluation of funds expended on fixed and liquid assets, can be determined from the following formula:

$$C = \frac{(K - O \cdot W \cdot P)}{a \cdot t \cdot f \cdot q}$$

where: C is prime cost of output unit, in rubles;

W is amount of operating expenditures for the entire period of exploitation of a deposit, on deducting amortization, in rubles ($W = a \cdot t \cdot V$);

a is yearly productivity of ore, in tons;

f is mean content of metal in ore, in percent;
 q is coefficient of recovery of metal on all stages of the technological process;
 γ is content of metal in finished production, in percent;
 O is remanent value of solid assets, in rubles;

The other symbols retain their previous connotations.

In the evaluation of a deposit and determination of unit rate of capital investments and prime costs of production, the comprehensiveness of utilization of raw material is of vital importance. Here, however, it is necessary to take into account the national-economic demand for the individual components and the balance of its satisfaction. For instance, the needs of the national economy for a number of rare components (vanadium, germanium, etc.) can be fully and less expensively satisfied by the recovery of these metals not from nonferrous-metal ores but from iron ores and by the coke-chemical industry. Therefore, when evaluating polymetallic deposits it is necessary to proceed not from the maximal but from the optimal recovery of the components contained in the raw material. The unit rate of capital investments during a comprehensive utilization of the raw material can be determined in terms of the value of the recovered components, per ton of raw material or per ruble of finished production, while the distribution of capital expenditures among the individual recovered components can be determined in proportion to their wholesale-price value.

Evaluation of Deposit, and the Designing and Planning of Construction of Ore Mines

The results of the evaluation of a deposit should be utilized in the designing and planning of detailed geologic-exploring activities and constructing individual enterprises of the mining industry.

The evaluation of deposits should be conducted in successive stages. The preliminary evaluation of the expediency of detailed exploration is conducted by geologists from the standpoint of the importance of a given mineral to the national economy, on the basis of the balance of the mineral raw material, content of metal, thickness of ore body and conditions of its occurrence, strength of ore, volume of detected reserves, material composition of ore -- factors determining the technology and economic indexes of the processing of the ore.

On the basis of these data, by analogy with the ex-

existing or planned enterprises, the following principal economic indexes are roughly determined: production costs, unit rate of capital expenditures, profitability, and construction period. A comparison of the indexes thus determined with the analogous indexes of existing pace-setting enterprises makes it possible to determine the value of the given deposit and the expediency of its detailed exploration.

The second stage of the evaluation consists in the compiling of long-range plans of development of individual branches of the mining industry. For this purpose, on the basis of more detailed calculations of all promising deposits of a given group of metals, it is necessary to determine the principal technical and economic indexes characterizing the economic effectiveness of the development of individual deposits or expansion of existing ore mines.

Such a comparative evaluation serves as the basis for compiling technical-economic reports which justify, on taking into account the long-range needs of the country, which deposits should be brought under exploitation and in what order of sequence.

The comparative evaluation of the effectiveness of development of individual deposits serves as the material for stipulating, in the long-range plans, the calendar periods of construction of individual mines and for drafting on this basis the appropriate project designs. In a Socialist economy the solution of the problem of the sequence of development of individual deposits may be affected not only by economic effectiveness but also by political, strategic and other considerations. However, such a solution should be decided upon on taking into account the material expenditures it will incur. "The criterion of effectiveness is not a regulator but an objective, well-founded and precise mechanism for the planning of the national economy, including also construction" (Bibl. 11).

The economic evaluation of individual deposits is not fixed -- it varies according to the needs of the country, technological progress, and discovery and exploration of deposits. Thus, considering the recently discovered possibilities for replacing lead by aluminum and plastics, there is no need to bring under exploitation the medium and small deposits of low-grade lead ores requiring a high unit rate of capital expenditures and involving high prime costs of the metal. Therefore, the evaluation of deposits cannot be episodic in character. It should be conducted continually and systematically in the process of the compilation of long-range and, particularly, current plans of development of the national economy.

The presence of long-range plans of sequential devel-

opment of individual deposits does not exclude the possibility and expediency of introducing proper revisions into these plans. However, this does not lessen the need for conducting such a comparative evaluation of deposits, which will ensure such a consecutiveness of their exploration and development as will yield the greatest effect to the national economy.

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